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DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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MINING METHODS AND COSTS AT GRANADA GOLD MINES (LTD.), ROUYN, QUEBEC

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MINING METHODS AND COSTS AT GRANADA GOLD MINES (LTD.), ROUYN, QUEBEC[1]

By R. L. Loofbourow[2]

INTRODUCTION

The adjacent towns of Rouyn and Noranda, centers of the Rouyn mining district of western Quebec, are 360 miles by air line almost due north of Toronto. Rouyn is reached from the main lines of the Canadian National Railways and the Canadian Pacific Railway at North Bay, Ontario, by the Temiskaming & Northern Ontario Railway and its subsidiary, the Nipissing Central Railway. Noranda is the southwestern terminus of a branch from the Canadian National Railway between Winnipeg and Quebec.

The Rouyn mining district is a part of a mineralized zone that has been traced for a distance of 140 miles from Kirkland Lake in Ontario eastward into the Province of Quebec to the Pascalis district.

From Rouyn the mine is 5 miles almost due south over a gravel road built by the company with assistance from the Province of Quebec.

ACKNOWLEDGMENTS

To R. C. Gamble and W. A. Gamble, respectively president and secretary-treasurer of Granada Gold Mines (Ltd.), acknowledgment is made for information used in the section on the history of the mine and the district, and for permission to present this paper for publication. Claude Ferguson, manager, E. W. Todd, consulting geologist, and Thomas Evely, mine captain, have helpfully criticised various sections of the paper in the course of their preparation. S. D. Cooke, accountant, assisted in compiling the cost data.

BRIEF HISTORY OF THE DISTRICT AND THE MINE

Attention was first called to the Rouyn mining district in the fall of 1922 by the staking of the Powell property 3 miles northwest of the Noranda mines. For a year and a half staking went on in a large way. The Granada Gold Mines property was staked in January, 1923, by R. C. Gamble and his associates. It was prospected by running traverses across the property at intervals of 500 feet and plotting all outcrops, contacts, and structure. Gold was first found in September, 1923, veins were prospected more in detail by trenching, and shaft sinking was begun in February, 1927. For two years, during which much underground work was done, the road was a trail impassible to motor vehicles. Equipment and supplies were sledded over snow roads in winter and dragged through the bush in any manner possible in summer. The surmounting of these difficulties by the men who brought in the necessary heavy equipment and supplies stirs the admiration of anyone familiar with the country.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6709."

2 - One of the consulting engineers, U. S. Bureau of Mines, and chief engineer, Granada Gold Mines (Ltd.), Rouyn, Quebec.

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Until June, 1930, operations consisted solely of exploration and development. At that time a 75-ton mill was built to treat ore already broken and blocked out. Two Diesel engine-generator units were brought in to supply power for mine and mill. In the late fall of 1931 a power line was built to the mine; hydroelectric power has been available since, making possible an increase in plant capacity from 75 to 105 tons per day.

GEOLOGY[3]

The district lies in the pre-Cambrian shield near the "height of land" dividing drainage between the Ottawa and St. Lawrence Rivers on the south and Hudson's Bay on the north; the altitude is about 1,000 feet. Drainage is ill defined and circuitous; low but steep ridges and knobby outcrops of the more resistant rocks give a low relief that breaks the general flatness of the country. Black spruce, poplar, birch, balsam, and tamarack, providing a fair grade of mine timber, cover thickly all but the most rocky parts of the terrain.

The bedrock of the region consists of regionally metamorphosed pre-Cambrian volcanics, sediments, and intrusives. The local country rock is interbedded greywacke and conglomerate of the Temiskaming series into which have been intruded irregular, fingering sills of feldspar-porphyry, with which mineralization is probably associated. In the vicinity of the veins the country rocks are generally carbonatized and in places bleached. The sediments have a marked schistosity which strikes roughly N. 70 W. and dips 45° to 55° to the nort... The conglomerate bowlders are strikingly elongated parallel to this schistosity. Also, roughly parallel to it, are the boundaries of the porphyry sills, except where these follow faults.

Gold has been found in five high-temperature[4] quartz veins which are roughly parallel to the schistosity of the sediments. These veins, for the most part, are in the sediments, but close to and in places walled by porphyry sills. The width, richness, and nature of the vein filling are quite erratic, but the veins do not appear to be influenced by a change of wall rock from sediments to porphyry or vice versa.

The universal and predominant vein mineral is quartz. It is in general more or less crushed, and well-formed crystals are only rarely seen. The color is in places quite gray and in others milky white. Carbonate is found in the veins and, with quartz, in stringers in the walls. Talc or sericite, chlorite, and tourmaline are common but always occur in small amounts. The sulphides pyrite, galena, sphalerite, arsenopyrite, and chalcopyrite, named in order of prominence, constitute a very small percentage of the vein filling. Gold occurs chiefly in the free state, and ranges in size from barely visible colors to quarter-ounce particles ramifying through quartz, generally near small seams and zones of small-scale brecciation.

The width of quartz in the veins is erratic, the quartz commonly but not always occurring in kidneys or beads, elongated horizontally. The wall rock envelops the lenses in tight shells as though it had flowed under high pressure. In other places the quartz is more continuous, possibly having been rigid enough to withstand kidneying. The greatest thickness of quartz so far encountered is 12 feet; some sections of the veins average 5 to 6 feet and others less. Gouge is almost completely absent, though the walls of the quartz are slickensided and there exist in the quartz itself numerous seams parallel to the walls on which there may be a very small amount of crushed country rock.

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3 - A more complete report of the geology of the mine is shortly to be published. See Hawley, J. E., Geology of Granada Gold Mine, Rouyn Township. Annual Report, Quebec Bureau of Mines, 1931.

4 - High temperature or hypothermal, judging by the common occurrence of small amounts of black tourmaline (schorlite). Lindgren, W., Mineral Deposits: McGraw-Hill Book Co., 3rd ed., 1928, p. 718, et. seq.

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Although the wall rock contains a small amount of gold not apparent to the eye,[5] only solid quartz and stringer zones have so far proved to be ore. Assay-values in the quartz are erratic because of the presence of coarse gold in shoots of high-grade ore. Lower-grade quartz generally contains enough gold to pay the costs of mining and milling and therefore is completely removed in the search for high-grade portions of the veins.

Apart from the common occurrence of coarse gold in richer sections of the veins, experience has shown that the nature and amount of fracturing are the best guides for visually determining whether or not quartz is of good grade. The "open," "live" appearance arising from the presence of fracture networks through the quartz, and seams in it paralleling the walls, indicates the possibility of high-grade ore, whereas the glassy or "tight" appearance caused by the lack of these structures is typical of poorer ground. This distinction, though not striking, is readily made by one familiar with this and other similar deposits.[6] The presence of sulphides has not proved to be a guide, nor does the width or color of quartz appear to have any relation to the occurrence of high-grade ore. The presence of much calcite or chlorite in the quartz is not a favorable indication.

Nine faults with individual displacements of more than 15 feet cut the veins within the area so far explored. Although not quite parallel to each other, these faults are nearly at right angles to the vein; their dip is vertical or steep. The drag of vein material into and along faults is evidence that at least part of the movement has been postmineral.

PHYSICAL CHARACTERISTICS OF ORE AND WALL ROCKS

Shrinkage stoping was the method first used in mining the ore, but this method has been abandoned for open stoping. Several factors having an important bearing on the choice of a mining method are enumerated below.

1.- The walls are strong, but the schistosity of the rock within 1 or 2 feet of the quartz makes the immediate hanging wall likely to slab in places if it is left unsupported over large areas. This gave no trouble while mining was in progress, since the broken ore furnished the needed support; it would have been unobjectionable during drawing were it not for the following consideration.

2.- The dip of the veins (45° to 55°) is not sufficiently steep to allow even drawing over a necessarily uneven footwall. After one-half or two-thirds of the ore had been drawn from the shrinkage stopes the rest remained as long tongues extending up between the chutes (20 to 25 feet apart) for as much as 100 feet into the stopes. Slabs then detaching from the hanging wall fell directly into the chutes, blocked them, and made extraction of the remaining ore difficult.

3.- Irregularity and sharp changes in thickness of the quartz necessitated, in shrinkage stopes, the breaking of a large amount of waste to make drawing possible.

4.- Because of the irregular tenor of the ore, open stoping is advantageous. Under this system, ore broken anywhere in the mine may be brought to the mill in a few days, and greater uniformity in the assay value of mill heads can be maintained.

5.- The length of a stope on any level is limited by the faults that bound the stope. Since these are nearly parallel, vertical, and at right angles to the strike of the vein, stoping areas are panels of ore extending up and down the dip of the vein and separated by similar, though generally narrower, panels of waste where wall rock meets wall rock along the

5 - Samples containing no vein quartz invariably show a trace of gold and many assay 20 to 40 cents per ton.

6 - Ferguson, H. G., "Gold Quartz Veins of the Alleghany District, California": U. S. Geological Survey Prof. Paper No. 172, 1932, p. 60, mentions this in the Alleghany district. It is well known to experienced miners in the Grass Valley district in California.

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faults. In that section of the mine where exploration and mining have been concentrated, the spacing of the major faults is such that panels of ore may be mined completely, the panels of waste between acting as pillars (fig. 1).

6.- The irregular, curved nature of the walls and the absence of gouge on either hanging or foot wall prevent resuing, which might otherwise be used in the thinner parts of the vein to lessen dilution. Where the quartz is 3½ feet or more thick, little waste is broken in stoping and dilution is slight.

7.- Sorting of waste in open, stulled stopes is possible because of the concentration of gold in the quartz, which is easily distinguished from wall rock by its appearance and its tendency to break fine. Sorting is impossible in shrinkage stopes.

PROSPECTING AND EXPLORATION

Because outcrops are few and vein croppings are easily eroded on account of the shearing that has taken place along veins, surface prospecting is difficult, generally involving trenching. Diamond drilling from surface and underground stations has proved valuable as a guide to structure. Good core is obtained of wall rock and quartz, but, because of the erratic nature of the latter with regard to both thickness and tenor, no estimates of width or value can be based on examination of any quartz which may be found; if no quartz is found, little is proved, owing to the prevalence of small and large faults. Deep-hole drilling with hammer drills has not been tried; it would be open to the same objections as diamond drilling and would in addition produce no core for visual inspection. Six-foot holes in either hanging wall or footwall, drilled from a drifting set-up, are frequently useful in locating the main body of quartz in a stringer zone.

The only adequate method of exploring a section of a vein is to crosscut to it, and to drift and raise on it. Crosscutting and drifting alone, even at short vertical intervals, are not sufficient because of the tendency of quartz lenses to elongate horizontally. The attitude of veins and major faults is fairly consistent, so far as exploration has revealed. In exploring under known ground it is possible to follow a vein through faults by careful observation of such indications as drag, attitude of walls encountered, and direction of movement on these walls, and by following the general idea of the structure as obtained by projection. Drifts and crosscuts are driven approximately 5 to 7 feet in cross section; they remain open indefinitely without timber and are used for haulage where ore is found.

SAMPLING

Five types of samples are taken from the mine: (1) Channel samples, weighing approximately 5 pounds, are cut daily with hammer and moil across the face of each drift. The channel is 2 to 3 inches wide and about ½ inch deep. If either or both walls carry stringers, they are sampled separately. (2) Samples are cut in a similar manner at 5-foot intervals across the backs of drifts. (3) Rib samples (waist-high horizontal cuts 5 feet or less long) were taken along all crosscuts on the upper levels and are taken at present where stringers are encountered in crosscutting. (4) Car samples of all ore from stopes and development workings are taken every other day. The weight of these samples averages about 20 pounds, varying with the amount of ore taken from the place sampled. (5) On occasion, samples are cut at 5-foot intervals across the vein in the backs of stopes.

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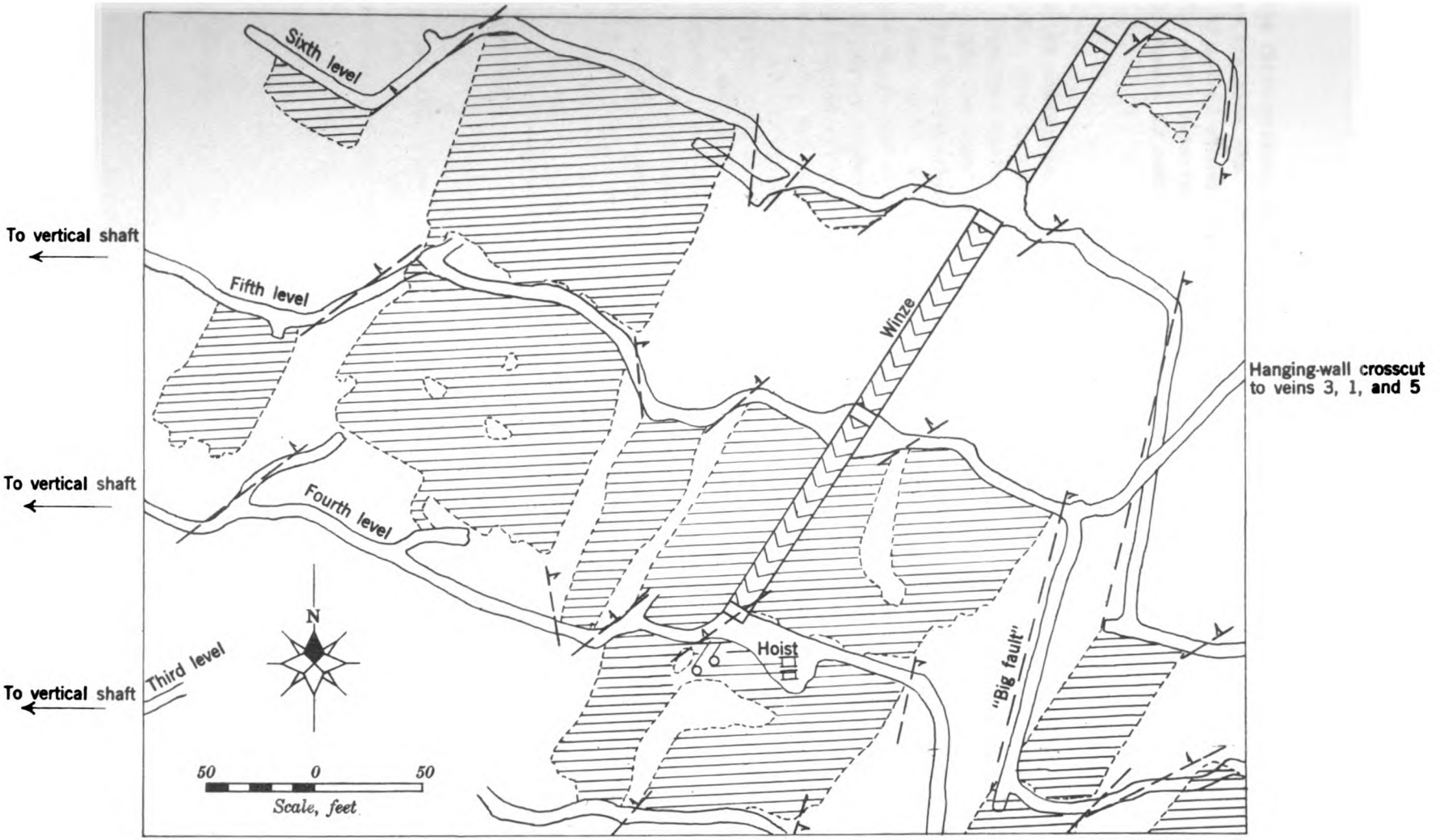

Figure 1.—Composite plan showing lateral workings and stopes (crosshatched) on part of No. 2 vein

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ESTIMATION OF TONNAGE BROKEN

On the first working day of each month, stopes and development workings are measured and plotted on the mine map. The tonnage of rock broken in stopes is estimated by multiplying the area of ground stoped (on the plane of the vein) by the stope width normal to the vein (average of width measurements), and dividing the product thus obtained by the tonnage factor, which equals 13 cubic feet per ton of rock in place.

DEVELOPMENT

From the standpoint of access, the mine may be considered in two divisions: (1) The upper levels, down to and including the fifth, are reached directly from a 690-foot, 2-compartment, vertical shaft collared on the outcrop of No. 1 vein and cutting the No. 2 vein at 530 feet. (2) The lower levels to the ninth (lowest at time of writing, October, 1932) are tributary to a 2-compartment winze collared on the fourth level 620 feet east of the vertical shaft and sunk on No. 2 vein. The fifth level serves as a main ore and supply haulageway between the winze and vertical shaft. On the fourth level the waste broken in development and that sorted by trammers on the lower levels is trammed to a point where it is dumped into old stopes between the fourth and fifth levels.

Since drifts and crosscuts serve the purpose both of exploration and development, the selection of the level interval is a compromise between the need for adequate exploration, the development of a sufficient number of points of attack, and economy in cost. Stations are cut every 125 feet in the vertical shaft and every 150 feet, inclined distance, in the winze. However, the sections of the mine that may be reached only by long crosscuts from the winze are not entered on each level but on every second or third level. For instance, the section of No. 2 vein east of the "big fault" is reached by a crosscut along this fault on the sixth and eighth levels but not on the seventh level. This section of the vein will be explored by a raise on the vein between the sixth and eighth levels; in order to maintain a number of points of attack, the vein may be cut by an intermediate drift (not, however, connected with the winze) so that either the upper or the lower section of the vein may be worked independently.

Each stope is served by one manway in which ladders, water and air pipes, and a wooden chute for steel are placed. After stoping is begun, a raise generally is driven to the level above for ventilation and for handling timber and steel into the stope. One or more ore passes, depending on the lateral extent of the stope, bring broken ore to the level for tramming.

Development Details

(1) Winze

Size of excavation:	7 by 15 feet.
Timber:	Round sills every 6 feet support ladders, pipes, and 4 by 6 inch stringers for rails. The back is unsupported except by an occasional stull. A heavy bulkhead, with a door to allow passage of a skip, protects men working in the bottom.
General method of driving:	The opening is carried full size.

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Development Details - Continued

(1) Winze - Continued

Machines and mountings: Two Ingersoll-Rand N 82 wet drifters are mounted on 6-foot columns with 2-foot bars.

Steel: One-inch, quarter-octagon, hollow steel with plain shank and double-taper cross bit is used in lengths of 2, 3, 4, and 5 feet; gage of starters is 2-1/8 inches, and 1/8 inch is dropped with each change; 60 to 100 pieces are used in drilling a round of 34 holes.

Round: A center wedge-cut round of 34 holes with six cut holes and six easers inclined toward them, breaks 2.12 feet per shift worked (4.24 feet per drilling shift).

Cycle of operations: Two shifts work each day, one sets up the machines, drills, and blasts; the next shift mucks out. Each shift has four men.

Explosives: 200 to 220 sticks of 1 1/8 by 8 inch, 40 per cent gelatin are used per round. Electric blasting caps, in delays up to 10, are used for detonation.

Mucking: Four men with round-point, D-handle shovels muck 35 to 40 tons per mucking shift (8.7 to 10 tons per man-shift) into a 1¼-ton skip.

The winze was started from the fifth level with a 1-ton skip drawn on a 3-foot-gage track by an air hoist. Sinking was interrupted while the sixth level was developed, but was resumed thereafter, and 300 feet of work was done. Ore mined from the sixth level during this period of sinking was hoisted in the winze during the drilling shift and at any time during the mucking shift when the skip could be used. Obviously the hoisting equipment in the winze was taxed to its limit and could not be expected to provide transport for development rock from the seventh and eighth levels in addition to the duties it then had. A plan devised by the mine captain, Thomas Evely, was put into effect. In addition to relieving the transportation problem in the winze without interrupting development of the two lower levels and mining on the sixth level, it made possible the direct disposal of waste into empty shrinkage stopes above the fifth level (fig. 2). The 3-foot-gage track was made into two tracks of 2-foot 4-inch gage, with 8 inches between tracks, by the addition of two rails, one 2 feet 4 inches west of the western rail of the original track and one between the two original rails and 2 feet 4 inches west of the eastern rail of the original track. The original track was used up to the day of the change. Above the sheave wheel, protected by a bulkhead, the two tracks were extended through an old stope to the fourth level where new sheaves and a double-drum electric hoist were installed. Skips and a 16-foot man-and-supply truck were built for the new tracks. The change was made by taking off the old skip, tearing out the old sheave wheel and the bulkhead which protected it, connecting the tracks through, putting on the new skips, and reversing the fishplate bolts in the west rail of the original track.

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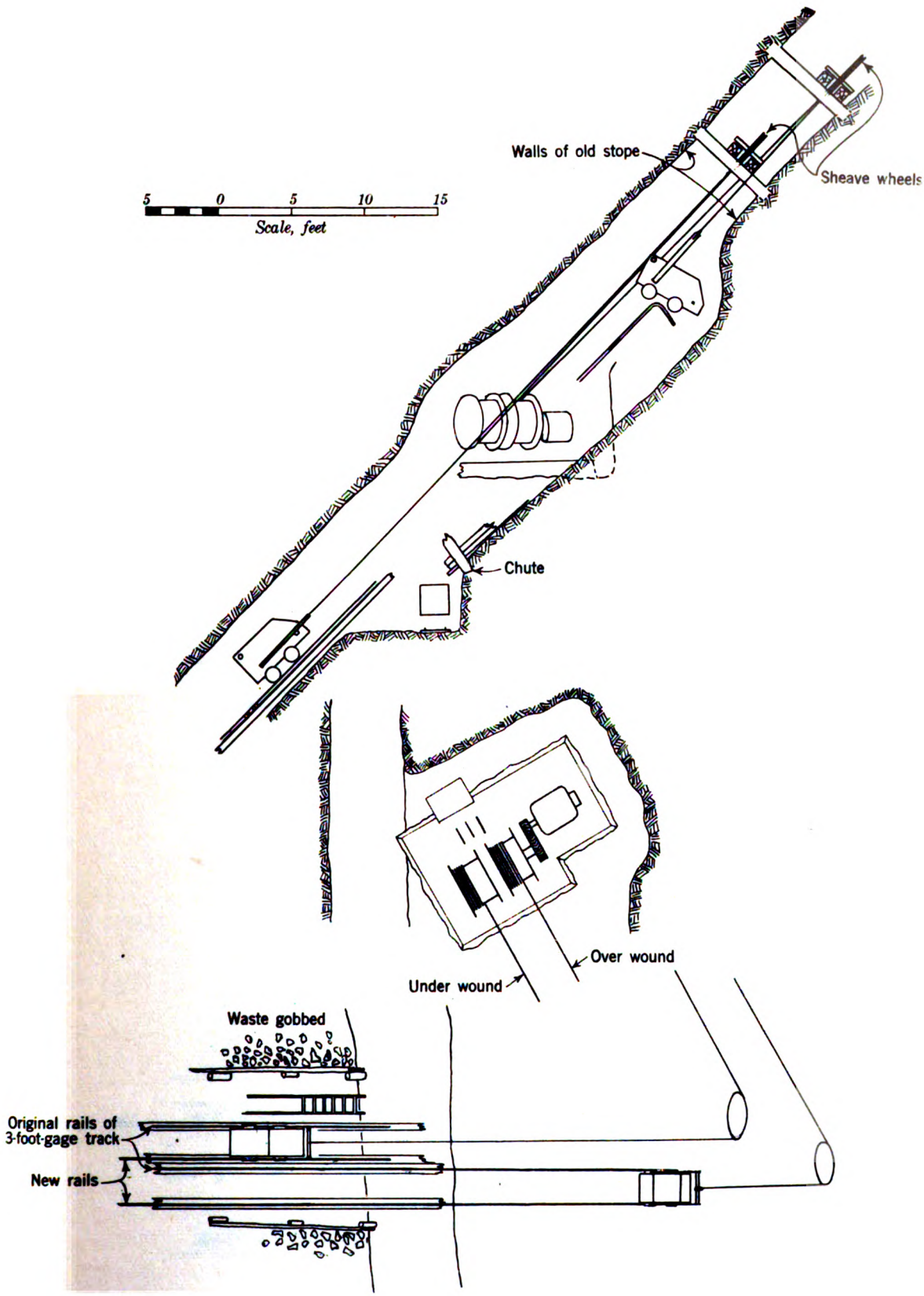

Figure 2.—Elevation and plan of hoist installation on fourth level

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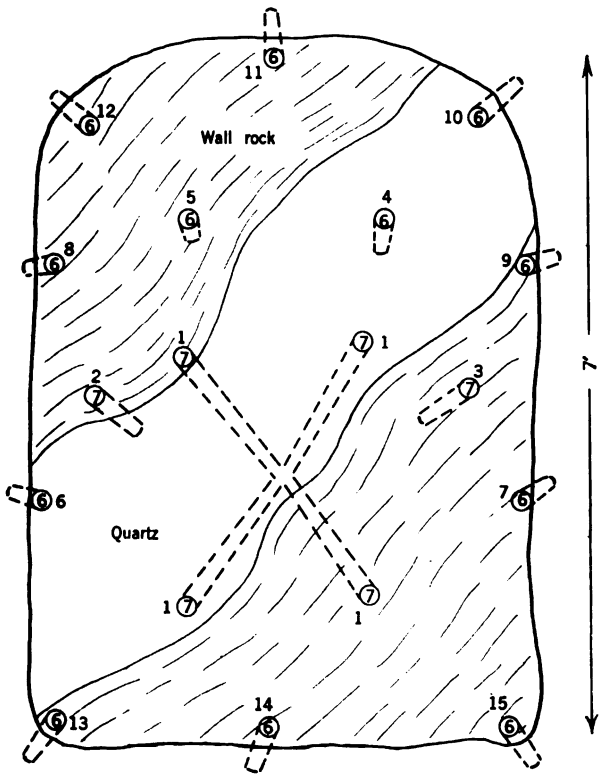
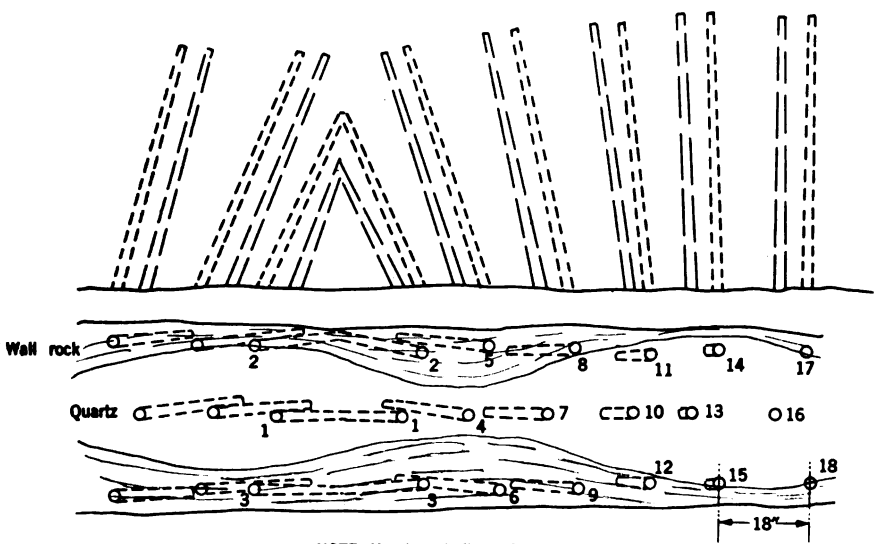


NOTE: Numbers in circles indicate sticks of powder.
Numbers alongside circles indicate firing order

DRIFT ROUND



NOTE: Numbers indicate firing order

STOPE ROUND

Figure 3.— Types of rounds used in blasting

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Noteworthy economy has been effected by placing the hoist in an old stope rather than in the footwall, thereby eliminating the necessity of crosscuts for access and for cablewyas, and greatly reducing the amount of excavation required for the hoist room itself. A total of 300 tons of rock was broken to make room for the hoist and was gobbed in the empty stope immediately below.

Development Details - Continued

(2) Drifts and Crosscuts

Size of excavation:	5 by 7 feet.
Timber:	None.
Machine and mountings:	Ingersoll-Rand N 82 wet drifter mounted on a 6-foot column with 18-inch bar.
Steel:	One-inch, quarter-octagon, hollow steel with plain shank and double-taper cross bit is used in lengths of 2, 3, 4, and 5 feet. Gage of starter is 2 1/8 inches and 1/8-inch is dropped with each change. Twelve sets (48 pieces) are, in general, sufficient to drill 18 holes 4 feet deep, but more may be needed if most of the holes are in quartz.
Round:	A pyramid cut of 18 holes is generally used (see fig. 3). Exceptions are: (1) Where a thick body of quartz is encountered in drifting, better footage is made by using a burned-cut round of 21 holes. The pyramid-cut round under this condition does not break nearly to the bottoms of the holes. (2) In crosscutting along a vertical or nearly vertical fault, a side draw-cut to the wall of the fault may be used.
Cycle of operations:	Two shifts are worked each day. On each a machine runner and helper muck back, set up, drill, and blast. One mucker mucks and trams and muck broken by the previous shift, and lays mucking sheets.
Explosives:	Approximately 120 sticks (1 1/8 by 8 inch) of 40 per cent gelatin are used per round. Caps and 6-foot fuse are made up on the surface and shellacked.
Mucking:	Mucker uses round-point D-handle shovel for cleaning up the track and a square point for mucking off sheets, of which three, 4 feet square, are used.

(3) Raises

Size of excavation:	4 to 5 feet by 7 to 15 feet.
Timber:	A stull is placed every 8 feet and lagging is nailed to it to form an ore pass on one side and a manway on the other.

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Development Details - Continued

(3) Raises - Continued

Machine: Ingersoll-Rand R 51 self-rotating, wet stoper.

Steel: One-inch, quarter-octagon, hollow steel with plain shanks and double-taper cross bit is used in lengths of 2, 3, 4, and 5 feet. Starting gage is 2 1/8 inches and 1/8 inch is dropped with each change. From 12 to 25 sets of steel are used to drill a round, depending on the size of the raise and the amount of quartz encountered.

Round: A center wedge-cut round or, if there is thick quartz, a burned-cut round is drilled. The number of holes may be from 14 to 30, depending on the size of the raise.

Cycle of operations: In a small raise a miner and helper drill and blast every shift; in a large raise two shifts are spent in drilling a round, or two machines do the work in one shift.

Explosives: 40 per cent gelatin, fuse, and caps are used, the quantity depending on the size of the raise.

MINING

The system of mining used has been referred to as a form of open stoping, although filling is a part of the operation. The writer considers this designation proper, since the primary object of filling is the ready disposal of waste and not the support of walls. Moreover, the supporting action of the fill in the case under discussion is not unlike the supporting action of the casual pillars left in some open stopes.

The procedure followed in stoping is not entirely uniform but may be generalized as follows: From the drift under a block to be stoped a 5-foot wedge-cut round is drilled in the vein material for the entire stope length. This is blasted, and a similar slice is removed by blasting another round drilled from the top of the broken rock. The broken rock is completely mucked out, leaving the back of the stope about 14 feet from the track level. Timbermen then place stulls about 6 feet above the track and spaced 8 feet apart, except that every 25 feet a 5-foot space is left and a chute is built. A tight lagging of poles is placed on the stulls to make a bulkhead. From this the back is drilled again, but before this slice is blasted, inclines of poles are built to deflect the broken rock to low shoveling platforms, or sollers, near the chutes. After the round is blasted, muckers remove the top lagging of the inclines, which allows them to throw waste behind the inclines onto the bulkhead while shoveling ore into the chutes. At this stage a raise is generally put up to the level above for ventilation and convenience in handling timber and steel into the stope. On completion of the raise, the vein material is stoped as before, except that miners build temporary stagings for drilling until the back is 30 to 50 feet above the bulkhead. A new bulkhead is then built along the length of the stope 6 feet from the back. If the stope width is less than 5 feet, room is made for muckers by blasting a row of holes in the footwall before the bulkhead is put in. Ore chutes are built up to this new mucking level, and waste is thrown through holes in the new bulkhead to fall on the old one. Inclines are not built

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here, since the broken rock funnels itself over the sollers and need not be cleaned up until the bulkhead is abandoned. Abandonment takes place when the back is 30 to 50 feet above the bulkhead, depending on the condition of the walls. The following tabulation shows the importance in a typical month, that of August, 1932, of the underground sorting and disposal of waste.

Handling of ore and waste underground, August, 1932

A. Of 4,822 tons handled (in both stoping and development) –
   1,503 tons (31.1%) were gobbed underground as waste (includes waste sorted and gobbed in stopes and development waste hoisted to fourth level and gobbed there).
   3,245 tons (67.4%) were hoisted as ore.
   74 tons (1.5%) were hoisted to surface as waste.
B. Of 2,975 tons handled in stoping –
   365 tons (12.3%) were sorted by muckers and gobbed in stoped as waste.
   156 tons (5.2%) were sorted by trammers as waste.

If the stope is long, a track may be laid on the mucking level and all ore may be shoveled into a small car and trammed to a central chute.

The muck being shoveled in stopes is thoroughly wetted down at the beginning of each shift to check the dust and to facilitate sorting.

The minimum stoping width is about 3½ feet. Where vein filling is less than this in thickness, the footwall is generally broken so that the hanging wall may be left intact and therefore rendered less likely to slab. All stringers are removed, but these are generally confined to a crushed zone within a foot or less of the main body of quartz.

Ingersoll-Rand R-51, self-rotating, wet stopers are used with one-inch, hollow, quarter-octagon steel with plain shanks and double-taper cross bits. Lengths are 2, 3, 4, 5, and 6 feet; the starting gage is 2 1/8 inches; 1/8 inch is dropped with each change. The average number of holes drilled per machine shift is 14; 14 sets of steel are used (70 pieces).

The round drilled in stoping is shown diagrammatically in Figure 3. Five sticks of 40 per cent gelatin are used in each hole. The firing order shown is followed unless the hanging wall is less firm than the footwall. In this case the hanging-wall hole of each vertical row is fired before the footwall hole. Holes are generally fired in groups of 28 – that is, after two drilling shifts rather than after each shift. The distance between the horizontal rows of holes is from 1 to 2½ feet, depending on stope width. If the quartz is more than 5 feet thick, four rows of holes are used.

UNDERGROUND TRANSPORTATION

Chute Gates

Rack-and-pinion gates without measuring pockets are used for loading skips in the winze. A short lip with vertical sideboards guides muck into the skip. The size of the gate opening, 20 by 20 inches, is ample to accommodate rock that has passed shovelers in stopes or drifts. Operation is rapid and sure; spillage is very small in amount. A guillotine-type gate and a stop-board gate are used on the fifth level to load cars of ore to be trammed from the winze to the vertical shaft. Of these, the guillotine-type gate is superior in that it may be efficiently operated by one man. Uppercut arc gates are used on ore chutes from stopes.

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Tramming

Up to the time of writing (October, 1932) all tramming has been done by hand. At present, however, an electric storage-battery locomotive is being placed in service on the fifth level to haul ore from the winze and stopes on veins 1 and 3 to the vertical shaft. It is thought that this will considerably decrease the transportation cost shown in Tables 1, 2, and 3.

Cars hold approximately 1 ton, and are the full rotating, end-dump type, equipped with roller bearings and run on 18-inch-gage track. The distance from the winze to the vertical shaft on the fifth level is 665 feet. Stopes on veins 1 and 3 are 200 to 400 feet farther from the vertical shaft. The distances trammed on the levels below the fifth are less.

Hoisting

In the winze, rock is hoisted in two 1¼-ton skips operated in balance. Rope speed is 450 feet per minute. For handling men and supplies at the beginning and end of each shift, one of the skips is removed by means of a vertical switch, and a 16-foot man-and-supply truck is substituted. Two pockets are provided on each winze station, one holding from 20 to 40 tons and the other from 50 to 70 tons. One pocket is for ore and the other for waste; they are used interchangeably, as the needs of development or mining require. Each pocket has two gates, one over each track. At the top of the winze is a 30-ton bin for storing waste to be gobbed in empty stopes between the fourth and fifth levels. On the fifth level is a 45-ton bin for ore to be trammed to the vertical shaft.

In the vertical shaft, ore is hoisted in cars on a counter-balanced cage, used also for men and supplies. The rope speed is 900 feet per minute. The cage is landed for loading and removing cars on four "chairs" made of 2-foot lengths of 1 by 4 inch flat iron hinged in a nearly vertical position to the posts immediately under each station so that the tops of the irons may be extended about 3 inches out into the shaft to engage the bottom of the cage. Chair irons are controlled by connecting rods actuated in turn by a lever attached to the shaft post so that the irons may be withdrawn or extended by the cage tender as he stands on the station. On leaving a station he locks the irons back from the center of the shaft.

PERCENTAGE OF EXTRACTION

Extraction of ore is not less than 98 per cent; the ore left is mostly in 3 to 4 foot level pillars that extend about half the distance across the back of each stope to leave a footing for tracks on the level above. Although occasional pillars are left in stopes, they are in lean parts of the vein. Dilution of quartz by waste arises in part from slight overbreakage in stoping wide portions of the veins but to a greater extent from the unavoidable breakage of waste where the veins are narrow or irregular in width. J. T. Willey, the mill superintendent, estimates that the ore milled contains from 35 to 50 per cent of wall rock.

COMPARISON OF METHODS

Under the heading, Physical characteristics of ore and wall rocks, are listed four objections to shrinkage stoping. Of these one of the most important is that sorting in stopes is impossible with the shrinkage method. Under this system as formerly practiced, trammers sorted waste from a 4-foot shoveling lip, inclined at 20°, over which the broken rock passed after coming through stop-board chute gates. This made loading slow and necessitated double handling of the sorted waste. Although at present trammers sort some waste, they merely supplement the sorting done in the stopes.

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The substitution of open stoping for shrinkage stoping has resulted in (1) a somewhat decreased cost of mining in spite of a more complicated transportation problem, and (2) in the delivery to the mill of cleaner ore.

WAGE AND CONTRACT SYSTEMS

The wage schedule is as follows:

	Per shift	
Shift boss	$5.50	(8 hours)
Miner in development	5.54	do.
Helper in development	4.74	do.
Miner in stope	5.04	do.
Helper in stope	4.24	do.
Mucker	4.24	do.
Trammer	4.24	do.
Hoistman	5.40	do.
Deckman	4.50	do.
Cageman	4.75	do.
Skip tender	4.24	do.
Timberman	5.04	do.
Timberman's helper	4.24	do.
Tool nipper	4.50	do.
Sampler and engineer's helper	4.50	(9 hours)

Contracts have been let for raising and for sinking the winze. Payment in both cases is for labor only, the company furnishing all supplies. In raises the contractors do all timbering, with the exception of the chute at the foot of the raise. In the winze all timbering is done by the company. Rates that have been paid are:

Raise.....$7.75 per foot to 4 men, two on a shift.
Winze.....15.00 per foot to 8 men, four on a shift.

Contractors earn somewhat more than men doing corresponding work on day's pay. The cost of raising has not been reduced by contracting the work at the above rate, but speed of raising has been greatly increased. The cost of sinking has been substantially decreased and the speed of work increased by contracting it. In general, contracting has greatly increased the regularity of operations and decreased time lost from incomplete work.

VENTILATION

At the time of writing (October, 1932) the mine has only one outlet to the surface, but a raise now being driven will shortly provide an airway and an emergency exit. Ventilation is now forced by blowing air between shifts through the compressed-air supply pipes to all working places in the mine. Valves are left open by miners going off shift and the compressor cut-out pressure is reduced to 25 pounds per square inch. Since the power cost is based solely on the peak load, which is not approached between shifts, the cost of this ventilation is negligible.

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ORGANIZATION

The organization of the company is as shown in the chart, Figure 4.

Table 1.- Summary of costs

Granada Gold Mines (Ltd.). July, 1932.

Ore hoisted during July: 3,104 tons.

Mining method: Open stoping.

Underground costs per ton of ore hoisted

	1 Labor	2 Super-vision	3 Compressed-air drills and steel	4 Power other than air	5 Explosives	6 Timber	7 Other supplies	8 Total
Development[1]	$0.423	$0.049	$0.254	-	$0.418	$0.020	-	$1.163
Mining	1.556	.087	.476	-	.606	.040	-	2.765
Transportation[2]	.764	.024	-	$0.063	-	-	$0.123	.974
General under-ground expense	.079	-	-	.013	-	-	.019	.111
Totals	2.823	0.160	0.730	0.076	1.024	0.059	0.142	5.014

1 - No segregation of costs between work in ore and work in waste.

2 - Includes surface hoist expense, and cost of decking and dumping ore and waste on surface, as well as cost of work actually done underground.

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Table 2.- Summary of costs in units of labor, power, and supplies

Granada Gold Mines (Ltd.). July, 1932.

Ore hoisted: 3,104 tons.

Mining method: Open stoping.

	Development	Mining (stoping)	Total
A. Labor (man-hours per ton of ore hoisted):			
Breaking (drilling and blasting)	0.748	0.809	1.557
Timbering	.153	.249	.402
Shoveling	.309	.617	.926
Haulage and hoisting	.494	.803	1.297
Supervision	.073	.145	.218
General	.045	.091	.136
Total labor underground	1.822	2.714	4.536
Average tons per man-shift underground	-	-	1.763
Labor, percentage of total cost	-	-	56.30
B. Power and supplies:			
Explosives (40% gelatin), pounds per ton	1.960	3.927	5.887
Timber, board feet per ton	.83	1.65	2.48
Total power, kilowatt hours	13.4	22.0	35.4
(1) Compressing	-	-	29.9
(2) Hoisting	-	-	4.08
(3) Pumping	-	-	1.42

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Table 3.- Detail of development costs in dollars and in units of labor, power, and supplies

	Sinking winze	Drifting and crosscutting	Raising
Size of excavation, feet	7 x 15	5 x 7	4 to 5 x 7 to 15
Timbered or not	Timbered	Not timbered	Timbered
Physical properties of rock	Uniformly hard; quite firm		
A. Labor (dollars per foot and man-hours per foot):			
Breaking (drilling and blasting)	$ 7.50	$3.01	$3.32
	7.558	4.70	5.19
Timbering	$ 1.65	-	$0.55
	2.85	-	0.95
Shoveling	$ 7.50	$1.25	-
	7.558	2.35	-
Haulage and hoisting	[1]$ 1.54	$1.56	$2.22
	[1]2.37	2.39	3.41
Supervision	$ 0.35	$0.19	$0.17
	0.35	0.19	0.17
Total labor	$18.54	$6.01	$6.26
	20.686	9.63	9.72
Feet advanced per 8-hour shift	2.117	3.40	3.08
B. Power and supplies (per foot):			
Explosives (40% gelatin)	$ 4.30	$2.60	$2.35
Pounds	25.73	15.67	14.15
Timber	$ 0.51	-	$0.17
Board feet	24.	-	8.
Power[2]	$ 1.55	$0.96	$1.07
Kilowatt hours	101.	63.	70.
Other supplies (caps, fuse)	[3]$ 1.63	$0.29	$0.30
Total power and supplies	$ 7.99	$3.85	$3.89
C. Labor, per cent of total cost	69.5	61.0	61.6
Power and supplies, per cent of total cost	30.5	39.0	38.4

1 - Comparatively low because during the period covered all rock broken in winze was gobbed underground as waste, whereas hauling and hoisting cost under drifting and crosscutting and raising includes cost of bringing ore to surface.

2 - For air compression only.

3 - Electric blasting caps were used in sinking.

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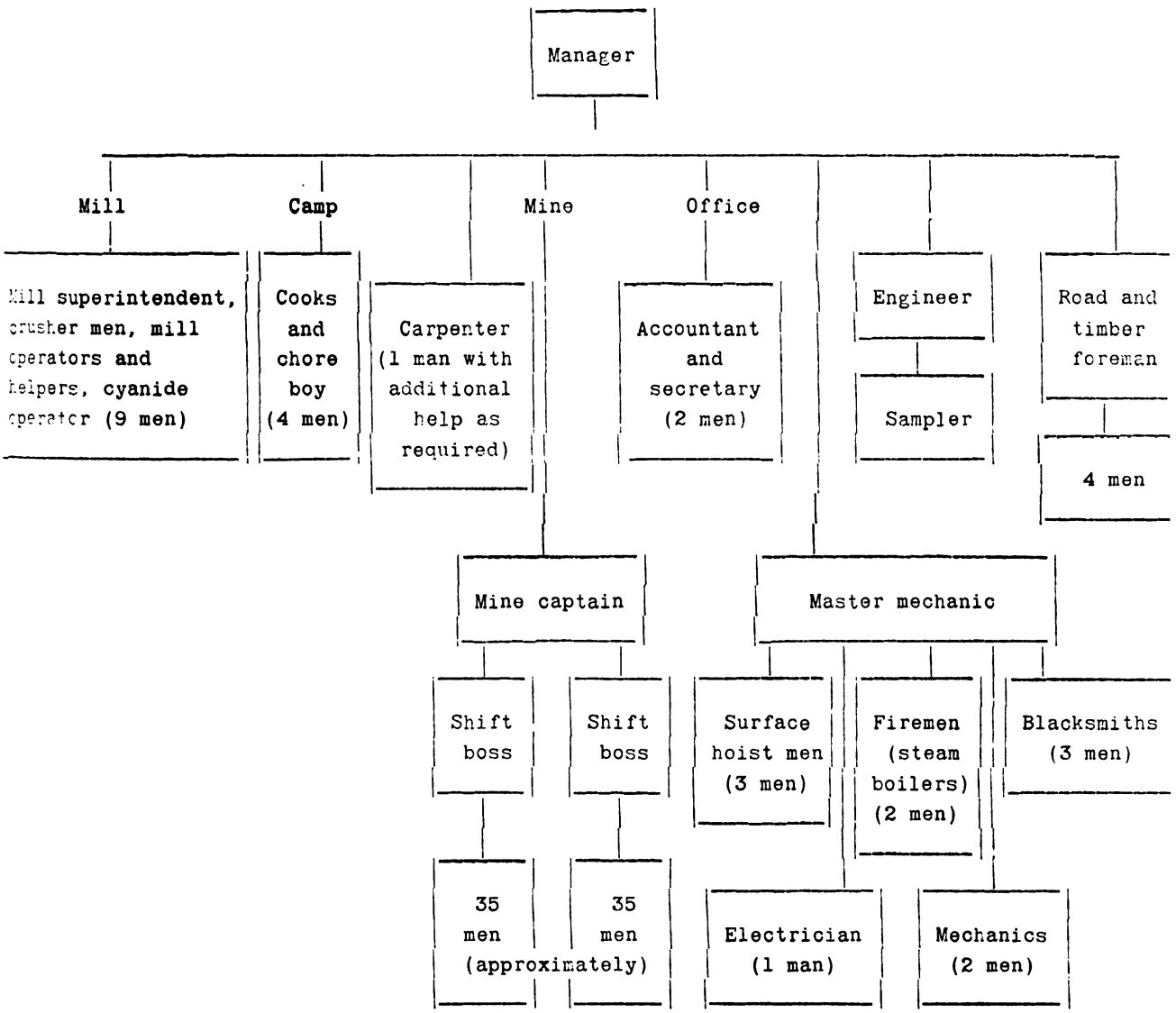

Figure 4.- Organization chart.

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